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Leaching processes for silver extraction

Phil Van Frank

THESIS

Silver Extraction

FRANK

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A thesis for the degree
of Mining Engineer
by
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The leaching processes for the extraction of silver are comparatively modern, being introduced in 1849. They belong to a period in which the metallurgist has brought to his aid knowledge derived from chemical research. Under favorable circumstances leaching processes possess many advantages over amalgamation or smelting. For successful use the ores treated for silver must not contain large per cents of lead, zinc, antimony or arsenic. The objections to these bodies will be named latter on. When silver is extracted by smelting or amalgamation, a costly plant is necessary which requires constant repair and in time wear out and be replaced, while on the other hand a plant for leaching is cheap. The roasting furnaces need not be of a very expensive type, the leaching tubs, being made of wood are cheap and for the most part low priced labor. The reagents used made be used again and again, as the loss is very small. These processes have one disadvantage, they must be constantly watched by an expert and continual assays must be made to see that there is no loss of

silver. The reactions are soon learned by the workers.

Leaching process may be divided into two classes; (1) Converting silver to chloride and dissolving (2) Converting the silver to a salt soluable in water. Under the first class fall Augustin process and Van Patera's process. Augustin converts the silver to chloride then leaches out the silver, chloride by a hot and strong solution of common salt. Van Patera leaches out the silver, chloride by hyposulphite of soda. Under the second class falls Ziervogel's process. In this process the silver is converted to silver sulphate and water is used for leaching.

On account of lead, antimony, and arsenic being frequently associated with silver, ores, wet methods are not extensively used and in most cases (when used) are applied to extraction of silver from a matt.

Augustin's Process.

This method is based on the fact that silver, chloride is soluble in a strong solution of common salt. From this solution the silver may be precipitated by copper, and the copper in turn ~~separated~~ precipitated by iron. This process is better applicable to such products ass matte and speiss than to the direct treament of ores, for the raw ores frequently contain substances which ~~frequently~~ interfere with a perfect chlorination of silver.

Copper mattes, containing from 50 to 70 per cent of copper and free from (or should contain only a small per cent of) lead, antimony, arsenic or zinc. If the last metalls are present a loss of silver will occur. In the latter part of this thesis, the objections to these metals will be explained. When mattes are poorer than 50% of Copper they are concentrated, if not done the residues will be to rich in silver.

Omitting the pulverizing and screening of the ore, this process consists of three steps.

I Roasting.

II Roasting with salt.

III Leaching and precipitation

The products obtained are cement silver, cement copper, residue containing 40 to 65 % of copper

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I. Roasting
II. Roasting with Salt
III. Leaching and precipitation

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and sulpate of iron. Spiess is more difficult to treat than copper matte since it does not give up the silver as readily as the matte.

Desilverization of Copper matt by Augustins Process

I. Roasting

The finely pulverized matt, consisting principally of sulphides of iron, copper, and silver, is roasted on the upper hearth of a double reverberatory for, about four hours; a low red heat being used. The ore is now moved to a double hearth, and roasted for two hours at a low temperature, the heat is now raised and the roasting continued for three additional hours, by this time the roasting should be completed. During the latter part of the roasting, samples are taken time to time until, when treated with hot water a slight blue solution is made, and gives a precipitation of silver up the addition of sodium, chloride. When this point is reached the first roasting is completed and the charge is removed from the furnace. During the roasting (1) the iron is converted to iron sulphate, (2) next copper sulphate and (3) sulphate of silver. The temperature, under which silver sulphate is formed is so high that most all the iron sulphate and a great part of the sulphate of copper are decomposed; so that at the end of the first roasting the mass consists of iron and copper oxyds, some undecomposed sulphides

and sulphate of silver.

II. Roasting with salt

As soon as the mass from the first roasting becomes cool it is finely ground and passed through a bolting sieve. About 300 pounds of this mass is charged into the same furnace in which the roasting was done; there it is mixed with from 3 to 5 per cent of salt. It is now roasted at a low temperature for about three hours. The sulphuric acid of the sulphates decomposes the salt, liberating chlorine, forming sodium sulphate and sulphuric acid. The free chlorine unites with the silver, nearly all of that metal and some of the iron and copper oxyds are converted to chlorides. When all of the salt has been decomposed the charge is removed from the furnace and taken to the leaching room.

III. Leaching and Precipitating

The leaching tubs hold about 400 pounds of roasted matt, each. They are 3 ft 9 in high, two feet eight inches diameter at the top and 2 ft 4 in at the bottom. Each of these tubs has a filtering apparatus on the bottom. First a wooden cross is placed on the bottom of the tubs upon which rests a perforated wooden bottom; over this is placed a layer of straw or twigs and above this is placed a piece of linen cloth and brought close against the

sides by means of a hook. The roasted
matt is placed on the cloth until the tub is
nearly full, now on top of the tub is placed
a perforated wooden lid and the tub is ready
for leaching. Brine (which has been heated) is
now run into the tubs. At first the stop cock
in the bottom of the tubs is opened to allow the
escape of vapor, it is then closed for a
quarter of an hour, and opened again. The
heated brine on coming in contact with the
silver chloride dissolves it and flows out of
the tubs saturated with this chloride. The
out flow (which is equal to the inflow) is run into
tanks in which the silver is precipitated as
cement silver. The copper need for precipitating
is placed in filters (like those arranged
in the leaching tubs) for a depth of six
inches. The silver solution is carried through
this; all the silver is precipitated; the desilverized
solution is run into tanks where the
copper is precipitated as cement copper
and used again for the precipitation of
silver. After the leaching has been completed
samples are taken from each tub
and assayed, if the desilveration has not
been sufficient, the residues must be
reroasted and leached. They contain from
40 to 65% of copper. The leaching liquors
are used again and again until they

become impure when they must be freed from
the sodium sulphate by crystallization.

The leaching is divided into two periods.
The first with concentrated solution until
a piece of clean copper do not become
crated with a precipitation of silver. This
period lasts from 20 to 22 hours.
The second period is the leaching with
clean water. When the washing water has
been sufficiently concentrated it is treated
for silver.

The precipitated silver is removed
at intervals of eight days, from the copper
on which it accumulates. Chloride of
lead and any sub chloride of silver is
dissolved by washing with water and dilute
hydrochloric acid. The silver is then melted
and refined.

Ziervogel's Process.

Ziervogels Process is based on the fact that silver sulphate is soluble in hot water and can be precipitated by copper after lixiviation. It is only applied to the extraction of silver from matt. The process as used in this country is divided into six steps.

(1) Crushing, calcining and roasting for silver sulphate
(2) Leaching the roasted matte.
(3) Precipitation of the silver.
(4) Washing and fusing the cement silver.
(5) Precipitating the copper.
(6) Refining the cement copper.

Crushing &c &c.

The matte is first broken up with sledges and then crushed so as to pass through a twelve to the inch screen. After screening, the matte is taken to the calciners. The furnaces are arranged with three hearths, a charge consists of one ton to each hearth. The third hearth is dark, this is necessary to prevent fusion, as their must be rapid oxidation at the lowest possible temperature, After remaining

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on the first hearth for 8 hours, the matt
is moved to the middle hearth and a
new charge put on the third hearth. The
middle hearth is ~~kept~~ at a dull red heat.
The charge remains on this hearth eight hours
after which time it is moved the the first hearth
(which is at a bright cherry red) and allowed
to remain 8 hours longer. The charge is now
drawn from the furnace and allowed to cool. A
small amount of sulphurous acid is given off
after the matt is taken from the furnace.
Three tons of matt are calcined in a day;
90 % of the sulphur is removed, the remain sulphur
is there as sulphides and sulphates. One man
can tend the operation, when he is not attending
the fire, he is stirring the charge. One and a
half cords of wood will calcine 3 tons of ore.

Roasting for Sulpate of Silver.

After cooling the calcined matt is ground
so as to pass through a 60-mesh screen. then
taken to a furnace to be roasted for sulphate
of silver. This furnace is called the fire cal-
ciner and has but an hearth, which is eleven
and a half feet long by 10 ft 6 in. wide and is
flat. The charge consists of 1600 pounds of
roasted crushed matt. Before introducing the
charge all the dampers are closed and the
hearth allowed to become dark. The fireplace
contains only embers, just enough to keep it

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hot. As soon as a charge is introduced it
is spread over the hearth making a
layer about three inches thick when spread
the damper is raised just a little, but no
fuel is added. In about an hour the charge
has a dull, blackish glow; but is red when
stirred. A small amount of fuel is now added
and the temperature raised until the charge
is at a dull red heat. The supply of air
comes from the bridge holes, the working door
and the grate; the fireplace door being closed.
At this period a maximum amount of iron sulphate
is formed and a small amount of copper sulphate; the
silver remains unchanged. Sulphuric acid fumes
are also given off by the decomposition of the
persulphate of iron, the charge increas in volume
and becomes spongy. From the second hour
the grate is kept full until the end of the
operation and the temperature kept as near
uniform as possible. At the end of three hours
the heat is at its highest point and silver sulphate
is formed and the charges becomes dry
no longer sticking to the rabble. The sulphate
of iron is decomped at the end of two
hours and the sulphate of copper at the
end of three. When the silver sulphate is formed
the charge is thoroughly stirred and all
lumps broken up. This stirring lasts an hour
so at the end of four hours the charge

is withdrawn. At the end of three hours assays are commenced and continued until the end of the operation. To make an assay the sample is thrown into cold. The heat of the sample is so great that the temperature of the water is raised to boiling. Whatever silver sulphate is present is dissolved but immediately precipitated by the suboxide of copper as metallic silver, and spangles are formed. At the end of the fourth hour all of the suboxide of copper has been converted to protoxide and no spangles are seen in the assay and the sulphate of silver is permanent. From 90 to 95 per cent of silver is converted to sulphate, the rest (being in a condition of arsenides, antimonides, or fine particles inclosed in lead sulphate) is unchanged. It would not be safe to decompose all of the sulphate of copper as some of the silver sulphate might be decomped and lost in the tailings, so when the spangles are no longer formed in the assay and a liquor of pale blue color is produced the charge is drawn, the furnace is cooled for another charge. only two charges are made a day and none at night as the operation is exceedingly delicate and requires careful watching. One furnace used ¾ of a cord of wood in two charges and is tended by one man.

Leaching the roasted matte.

After remaining on the cooling floor for twelve hours the roasted matt is moved to the leaching tubs. These tubs are 3 ft high, 3 ft diameter at the top and 2½ ft. at the bottom. They have a double bottom pierced with holes and covered with a cloth for a filter. The roasted matt is put into these tubs and the leaching done with boiling water. The tubs are kept full and the discharge flows into a series of tanks below. The time required for leaching varies with the richness of the matt; generally about an hour is required for every hundred ounces of silver contained in the matt. During the first of the process the mass in the tubs is light but in about an hour it settles and water does no flow through so freely. All the sulphate of copper is dissolved in the first part of the operation. The residues contain all of the gold and what silver was not converted to sulphate, they averaged about 25 ounces of silver per ton. As soon as the discharge from the tubs does not show any trace of silver with salt the leaching is stopped and the residues removed.

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Precipitating the Silver

The water (charged with silver and copper) from the leaching tubs runs into a set of

tanks in which plates of copper are arranged. The plates are so arranged that as the water flows through the vats it is exposed to 100000 square inches of copper. The vats are kept covered but at the end of each week the covers are removed and the copper shaken and removed, the cement silver falls to the bottom. When the silver has settled the copper solution ~~falls~~ is drawn of and run into tanks for the precipitation of copper and the silver is washed to remove all traces of copper. The amount of copper dissolved is equivalent to the amount of sulphuric acid set free from it combination with the silver.

Fusing the Cement Silver

The cement silver is removed from the precipitating vats and washed with dilute sulphuric acid (1 part acid to 100 parts of water) after thoroughly washing with the dilute acid, cold water is used for washing. The silver is then dried and melted in graphite crucibles. It is cast into bars and is from 999 to 999.5 fine. The time of washing is about three hours for 3000 ounces of cement silver.

As the rest of the process relates to the precipitation of copper and refining of the same no mention will be made of that.

Van Patera's Process.

Dr Percy in 1848 suggested the use of hyposulphite of soda as a leaching agent. Van Patera introduced the process in 1858. It is based on the fact that silver chloride is readily soluble in solutions of sodium or calcium hyposulphite. The silver can then be precipitated from these solutions by an alkaline sulphate with a regeneration of the hyposulphite solution. In this process the reactions are exceedingly delicate and the process need the careful attention of all employed. On account of the great variety of products to which it can be applied this process is likely to assume great importance in the production of silver. The steps are

I Crushing the ore
II Dry the ore
III Roasting the ore
IV Leaching out base metals
V Leaching with hyposulphite of soda
VI Precipitating silver with calcium sulphide
VII Roasting sulphide of silver and melting for bullion.

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I Crushing the ore.

The ore is crushed so as to pass through a 15 or 20 mesh screen. Thirty mesh screens were fist used, but is has been learned that there is no necessity in doing so. The ore roasts better, and leaches better with the coarser ore. With a coarse screen more ore can be crushed, and there are no fine particles to clog the filter. The size of screen for an ore can only be determined by trial. Ores which are apparently the same act differently in leaching. The ore should be crushed just as coarse as is consistent with perfect formation of ~~roast~~ chlorides in the roaster. Improper crushing easily translates itself to a loss of silver and longer time for leaching. From the crusher the ore is taken to the drier.

II Drying the Ore

The drying may be dried on a drying floor or in a revolving furnace called a "drier" The drying floors are made by passing the waste heat through horizontal flues with an iron covering. The ore is spread on this floor and occasionally stirred until dry. The "drier" does away with the stirring. It is simply a revolving ~~cast~~ wrought iron cylinder 20 ft long by 4 ft diameter. The ore is automatically fed at one end, and dumped into cars at the

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other. "Driers" generally have a seperate fire place but this is unnecessary, as the heat and flame from the "roaster" could be utilized. The "drier" is more convenient than the drying floor. After drying the ore is again screened, this time through a 15 mesh screen, and allowed to go down a shoot to the crusher. In the shoot are arranged magnets that remove all pieces of broken picks, drills &c &c.

III. Roasting Ore

The roasting is carried on gently at first and gradually raised until low red heat is obtained, then steam under low pressure is introduced into the furnace to drive of the last traces of sulphur, arsenic, and antimony, and also decompose all volatile base metal chlorides (The nascent chlorine thus give off acts very energetically on the silver. The ore must be constantly stirred. When lead is present the roasting must be done at low temperature (to avoid fusing) and the lead must be converted to chloride

The roasting takes from 8 to 10 hours. When completed the ore is removed from the furnace and left in heaps for 6 or 7 hours. Chlorine still acts in the heaps so nothing is lost by leaving them so long. From the heaps the ore is taken to the cooling floor and spread and allowed to cool until it will produce no change of temperature

of the water in the tubs. When this point is reached it is taken to the tubs and leached.

A considerable amount of silver chloride is volatalized in the roaster therefore dust chambers are made to collect this. When enough is collected it is mixed with the ore charged in the roasting furnace.

As yet I have said nothing about adding the salt for the reason that there is some discussion as to the proper time. In Europe a mixing machine is used to mix the salt and ore just before charging the roasting furnace. In this country when rolls are used the salt is added to the ore just before crushing and by the time it gets to the roaster it is thoroughly mixed. From 2 ½ to 10 per cent of salt is used, depending on the ore. Dr. Egleston says that if the amount of salt used was reduced 50% and the ore left longer in heaps that there would be a gain of 15 to 20% in chlorination. Mr. Stetefeldt thinks it best to use an access of salt.

IV. Leaching out base metals

The leaching tubs are filled to about 3 inches from the top and special care is taken not to disturb the ore after once in the tubs. Simply pushing a stick into the ore may delay leaching several hours.

When the ore contains little base metal the

leaching is done with cold water but when
base metals are present, first hot then cold
water is used. If hot water is used the leaching
is done from the bottom. This method allows the
water to percolate through the charge more
easily. The water is allowed to run until the
ore is covered, then the supply is cut off and
then allowed to remain ~~some~~ a short time. Now the
discharge pipe is opened and the water is
drawn off and the leaching water added
at the same time. If any silver is dissolved
by the excess of salt it forms a crust on top
of the ore and is carefully removed for subsequent
treatment. When lead is present in large quantities
the hot water must be introduced from
to top and the lead chloride washed out. The
dissolved chloride of silver is then precipitated
in the tanks that catch the leaching water.
The leaching is continued until the discharged
water give no precipitate with calcium sulphide.
The ore in the tubs must be cooled before
leaching with hyposulphite of soda.

V Leaching with Hyposulphite of Soda

The ores remains in the same tubs as for
the proceeding process and the hyposulphite
is introduced from the top. The strength of this
solution depends upon the richness of the ore.
When the ore is poor in base metals and rich
in silver the hyposulphite may be strong and

and a little warm but when the ore is
rich in base metals and poor in ~~the~~ silver the
hyposulphite must be cold and dilute.
When the leaching with water is poorly done and
sulphate of soda and sodium chloride are left
in the tubs the hyposulphite becomes impure.
It is then regenerated by spreading wood ashes
over the charge and Setting the hyposulphite
pass through these. The alkalies in the ashes
take up the sulphates and chlorides in the
reagent. By the use of calcium sulphide as
a precipitating reagent, hyposulphite of soda
is gradually converted to hyposulphite of
calcium. This is no objection for hyposulphite of
calcium dissolves gold as well as silver.

The leaching is continued until no precipitate
is produced by calcium sulphide. The time of
charging, leaching and discharging the ore takes from 18
to 48 hours, depending on the richness of the ore
and the manner of roasting. The reaction
during leaching is

$$2AgCl + 2\,Na_2S_2O_3 + 5H_2O = 2NaCl + 2NaAgS_2O_3 + 5H_2O$$

The silver solution has been run into separate
tanks from the base metal solutions and is
now ready for the precipitation of the silver.

The tailings in the tubs are not immediately
removed, as soon as the reagent quits
dissolving silver, Samples are taken from each
tub an assayed. If the samples assay over

six ounces per ton, the tailings are taken back to the furnace and worked over. From fifty to sixty tons of ore can be leached a day by five men and 24 tubs. Three of the men attend to the charging and discharging of the ore and two attend to the leaching

VI Precipitating the Silver

In this step is where this process has an advantage of those proceeding, the leaching agent is regenerated. The action is

$$2\,NaAgS_2O_3+2NaCl+10H_2O+CaS =$$
$$Ag_2S+2Na_2S_2O_3+CaCl_2+10H_2O$$

The calcium sulphide is added to the silver solutions until no further precipitation takes place. The solutions are then agitated for a few minutes and allowed to settle. The clear liquid is syphoned off and forced back to the tank from which all the hyposulphite runs. Care must be taken not to add an excess of sulphides as only that portion which precipitates silver is converted to hyposulphite of lime and besides silver would be precipitated in the tubs by this excess. Therefore the liquor must be neutral after all the silver is precipitated.

This process of precipitation is carried on for about ten weeks. At this time the

silver sulphide must be removed. This is done by first stirring up the sulphide until it is in the state of thin mud. It is then drawn off through the bottom and filtered and dried. When thoroughly dry it is removed from the filters and ~~s~~melted.

VII Roasting and Melting Silver Sulphide

If this is done in a reverberatory the process must be carried on slowly at first, only enough heat to burn of the sulphur. The roasting is carried on until nearly all of the ~~heat~~ sulphur is driven off, then the heat is raised as high as possible without melting. After roasting, the mass is nearly all metallic silver. It is mixed with lead and cupelled and cast into bricks.
The litharges are rich in silver and are reduced to lead for the next cupellation. This produces a bullion between 800 and 900 fine.

When the silver is treated in crucibles the roasted silver is mixed with iron and borax and fused. The iron sulphide formed comes to the top and is skimmed. The bullion produced this way is from 940 to 960 fine.

The purity of bullion depends largely on the amount of base metals present in the ore in the beginning of the process.

Most of the lead found in the ore is found in the bullion. The reason is, that none of the lead sulphate was leached out by water but was dissolved by the hyposulphite and precipitated with the silver. If not for lead this process would give a bullion as near pure as any of the lixiviation processes.

Russell's Modification

Mr. Russell, of Park City, Utah has patented a process nearly like the foregoing. His process has nine steps.

I. Crushing the ore
II. Drying the ore
III. Roasting with salt
IV. Leaching with water
V. Leaching with hyposulphite of soda
VI. Leaching with a double salt of cuprous hyposulphite and sodium hyposulphite
VII. Precipitating lead as carbonate
VIII. Precipitating silver with sodium sulphide
IX. Roasting and Melting fin Bullion

Russell based his process on the facts that metallic silver, silver sulphide and compounds of silver with antimony & arsenic are more soluable in the double salt than in hyposulphite of soda and that lead carbonate is insoluable in hyposulphite of soda. The only difference

between the two process is that in leaching
out the silver and in precipitating
lead. Russell first leaches with hyposulphite
of soda until nearly all of the
silver is dissolved then completes the
leaching with the double salt. He claims
that the leaching is more perfect and
more quickly done. Now he precipitates
the lead from the silver by soda ash. The
lead carbonate settles rapidly, leaving a
clear solution to be decanted into the
silver precipitating tanks. The silver is
precipitated by means of sodium sulphide.
If calcium sulphide was used it would
enter the leaching solution and would
be precipitated with the lead.

There can be no doubt that Russell's
method is good and that he frees his
silver of lead by taking a little longer
time. This method does not need such a careful
roasting as with other methods, on account
of the easy solubility of silver sulphate and
metallic silver in the double salt solution.

This method is now hardly a year old, but
if acts as is claimed will be one of the
principal process of the future, it is not
in practical operation on a large scale
as yet. The following experiments taken from
Mr. Stetefeldt's paper on this process show

a few things claimed by Russell.

Lixiviation tests with roasted ores from various mills.

Furnace used	Name of Mill	Ordinary Solution	Russell Sol.
Howell	Alice, Montana	90.0%	93.7%
Bruckner	Custer, Idaho	88.5%	91.5%
Stetefeldt	Lexington, Montana	92.4%	93.6%
Stetefeldt	Manhattan	94.1%	94.6%
Howell	Black Warrior	94.0%	95.2%
Bruckner	Bertrand	89.5%	95.1%

The difference in the percentage of silver extracted by the two solutions depends largely on percentage to which the chlorination has been carried.
When the chlorination is carried to a high per cent the difference between the powers of the two solutions is not as great as when the per cent of converted silver is less.

The following tables shows the solubility of silver in an ore subjected to an oxidizing roasting.

Experiment	{% silver extracted by} {Hyposulphite Sol }	{% silver extracted} {by Russels Sol }
No 1	30.5	64.1
No 2	23.9	65.7
No 3	23.2	47.7
No 4	24.1	62.9
No 5	23.1	67.3

In experiments No's 1, 2 and 3 ~~and 4~~ and extra fire was used. In experiment No 3 the admission of air was reduced and in No 4 the charge was not red hot when determined. These experiments

prove that should the chloridizing roasting
be imperfectly done and that oxide of silver
is in the roasted charge, that Russels Sol.
will extract more of the silver than can be
extracted by the hyposulphite solution. If
it is possible to extract silver by lixiviation
without using salt in roasting, not only the
~~expensive~~ expense of the salt is saved, but also
the cost of drying and crushing it and the
necessary for the purposes. For localities where
salt is very costly and ores of a low grade,
a decrease of the yield of silver, (of several per cent)
would still leave a balance in favor of
oxidizing roasting and the use of Russels
Solution.

Comparison of Processes.

Comparing the leaching processes with smelting and amalgamation processes, we see that leaching is a great deal the cheapest. The leaching plant is exceedingly cheap, the roasting furnace need not be an expensive one, the tubs, being made of wood, are cheap and easily repaired and lastly the reagent used is cheap, from the fact that it can be used again and again. The plant necessary to carry on smelting processes is costly, needs expense repairs and the ore treated goes through a large number of different steps. The same may be said in the ~~almag~~ amalgamation plant; only the wear and tear on an amalgamation plant is great than on a smelting plant and besides the mercury used is expensive and the loss is considerable. (Another advantage that leaching processes have over any other is that poorer ores may be treated.)

Comparing leaching processes with each other there is little difference as to cost. The main difference is in the amount of silver extracted.

The application of Augustin's or Ziervogel's processes is limited to copper matts and where the Ziervogel process is used the residues are treated by Augustins process. Ziervogel's method is the simplest of any but the roast-

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must be carefully done and requires the
labor of men specially trained for the work.

To be secessfully applied the matt used
in either of these processes should contain no
lead sulphide, for in roasting, the lead sulphide
is converted to lead sulphate and cause the
mass to sinter. Sulphide of arsenic or antimony
should not be present for, in the Ziervogel roasting,
the bodies in presence of silver sulphate are
converted to arsenate and antimonate of
silver, both insoluble in water. The same
salts are formed in Augustin's roasting crucibles
a large amount of salt is added. Zinceforous
matts must be freed of the zinc by concentration
smelting in reverberatory furnaces. If zinc sulphide
is left in the matte zinc sulphate would be
formed and it takes a high temperature
to break this up. But at high temperatures
the matt softens by commencing to melt, thus
causing richer tailings and also a loss
of silver by volatilization. Copper dioxide leads
to the formation of metallic silver. In both
processes complete control of temperature
must be had. The best field for the processes
is at Mansfield, Germany. The matte there
is pure and contains 70 to 72% copper.

Van Patera's has great advantages
over Augustin's and Ziervogels
from the fact that it is not
limited to matts but may be used

for treating a great variety of ores, especially those containing base metals. At present Van Patera's process or its modification seem to be the methods for poor ores, especially those containing base metals. It can be applied to any low grade ore, excepting those running high in lead and when an ore contains a large amount of lead it should be smelted and no other process tried. By using hyposulphite of calcium (this process) instead of hyposulphite of soda, this process may be used for the extraction of gold.

All of the processes require careful watching and continual assays must be made to see that there is no loss of silver or reagents. In Van Patera's process the loss of the leaching agent is quit an item; but if carefully done the increase of silver will pay for the loss of sodium hyposulphite.

The following table shows the cost of treating one hundred pounds of material by the three principal processes.

Place	Process	% silver in material	Cost	% silver loss
Mansfield	Augustin	.333	$ .99	.059
"	Ziervogel	.333	$1.68	.029
Bertrand Mill	Van Patera	.102	$ .45	.013

Remarks on the roasting of ores.

As the success of any of the leaching processes depends upon the roasting of the ore, a thorough description of the process appears necessary. It is impossible to carry on a rational and correct roasting according to the quality of the ore, or in regard to any special object unless the chemicall actions that take place are regarded and understood.

The principal object in roasting must be first the production of sulphates. The iron pyrites and other sulphides when red hot and acted upon by the oxygen of the air change to sulphates by oxydation of the sulphur. The sulphuric acid cannot unit with the metal in the sulphide until the sulphur is driven off and and oxyd formed, therefore it part of the sulphuric acid goes to the oxyd and make a sulphate and sulphurous acid. In roasting an argentiferous copper matt, sulphate of iron is first formed, then sulphate of copper and finally sulphate of silver. The formation of silver sulphate requires such a high temperature that the copper and iron sulphates are decomposed into oxyds and sulphuric

escapes leaving the silver sulphate unchanged. As soon as the iron and copper sulphates are decomposed, the roasting in the Ziervogel Process is complete but in the other processes (when salt is used) the chemical action is a little different. The iron and copper are converted to sulphates at a low heat the salt remaining is different. But as soon as these sulphates are formed, and no more sulphurous acid is given off, the heat is increased and the decomposition of the salt begins. The decomposition of the salt is preformed in two different ways:

I. The sulphates of iron, copper &c emit the sulphuric acid in vapor, which, in contact with the salt, forms sulphate of soda, setting the chlorine free. One part of the oxygen of the sulphuric acid goes to the sodium in the salt making sodium oxide which uniting with another part of the sulphic acid, forms sodium sulphate while the freed chlorine combines with the metals and sulphides in such a way, that one part of the chlorine combines with the sulphur making volatile chloride of sulphur and the other part unites with the metal.

II. The other way of decomposition of salt differs in its result by not emitting chlorine

gas but forming chlorides during the act of decomposition. The sulphate in contact with salt enter into an exchange of compounds. The sulphuric acid combines with the soda of the salt to make sodium sulphate; the metal of the sulphate combines with the chlorine of the salt making a chloride. The chlorination of the metals is performed by the direct action of of chlorine on the metals and sulphides and by contact of salt with sulphates. Hydrochloric acid is also made during the roasting when water vapors enter into the porous mass of the ore.

When an ore is to be roasted with salt it must be examined not only in regard to the quality and quantity of sulphides, but also in regard to the earth matters accompanying the ore. If there is a great deal of lime in the ore, it absorbs the sulphuric making calcium sulphate and remains in this condition during the roasting without being converted. On this account calcareous ores require as much more sulphides as is necessary to convert all the lime to sulphate.

If the ores to be roasted contain a large amount of sulphides, the roasting must

be done without salt until the greater part
of the sulphur is driven off; if not the
ore will cake and cause an imperfect roasting.
Ores with small per cents of sulphur do not
need as much attention during roasting and roast
better than ores high in sulphur.

Losses during roasting:--: In the Ziervogel process
the loss of silver is greatest when there is a large
amount of metallic silver in the ore, or when it is
produced, during roasting and when the mass
is in a loose condition, allowing the air to permeate
more freely. The loss increases when the temperature
must be kept high. In roasting without salt, silver
is not only lost by being carried off by the draught
but also by the conversion of silver to silver oxide,
which is volatile. If not combined with antimony
it will be deposited in metallic form as it losses
its oxygen at lower temperatures.

In Augustin's and Van Patera's methods
the loss of silver during washing
is less than in an oxidizing
roasting, provided there is no large amount of
base metals are present. The losses in roasting,
(oxidizing and chloridizing) vary according to the
impurities and run from 5% to 15% the average
loss is 8%.

Respectfully Submitted,
Phil Van Frank